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ABSTRACTS OF RECENT ARTICLES ON MINE SUPPORT

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W. R. CRANE

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ABSTRACTS OF RECENT ARTICLES ON MINE SUPPORT[1]

By W. R. Crane[2]

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RECENT PRACTICE IN SUPPORT IN MINES

Recent papers on support of roof and top rock in mines representing present practice in so far as it has been described by engineers in all of the principal mining countries, cover a relatively wide range of topics. The papers abstracted herein include such subjects as the testing of materials of support, failure and movement of rock above workings, effect of excessive pressures on rock masses, and the application in mines of timber, metal, concrete, packing, and filling support.

It is not within the scope of this paper to present a logical and orderly discussion of the elements of support and their application to conditions influenced by the material worked or the method of mining. Only the essential facts of a paper appear in the abstracts, the idea being to give a fairly clear picture of the behavior of various types of support under known conditions and to put into concise form important information concerning present practice, modifications of practice, and the trend toward new practice. With such valuable facts from others' experience made readily available, it is hoped that the engineer with particular and often trying problems may obtain suggestions that will be immediately helpful.

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GENERAL CONSIDERATIONS

Strength of Materials. Metal Ind., London, April 15, 1932, pp. 420-421. Tensile tests for brittle materials are of little value owing to the fact that there can be very little plastic readjustment, and it has been found very difficult to secure axial tension and uniform stress distribution for such materials. In the case of wood, it is so difficult to devise a test piece which will fail by tension rather than by shearing along the grain that the tensile test is rarely used.

The ultimate compressive strength of ductile materials is impossible to determine, but is sometimes reported by an arbitrary measurement of deformation. For brittle materials compressive strength is a fairly definite test result, although it probably represents destruction by shearing action, or by lateral strain, rather than by direct compression. It is probably a good index of service strength of brittle materials under compressive loads.

The cross-breaking strength can be obtained only for brittle materials and is correlated with tensile strength, but is frequently much higher than tensile strength tests, due probably to a slight flowage.

Further investigation of the physical properties of coal-measure rocks and experimental work on the development of fractures. Phillips, D. W., Trans. Inst. Min. Eng., London, vol. 82, pt. 5, pp. 432-450. The author discusses the behavior of rocks under varying conditions such as: (1) The effect of placing beams under a transverse load, (2) under compression tests, and (3) the production of experimental fractures.

The crushing strengths of coal-measure rocks are given and range from 4,000 to 20,000 pounds per square inch for sandstones and from 3,400 to 11,000 pounds per square inch for shales, which shows rather a remarkable variation in each.

The behavior of the roof, coal, and floor materials under forces operating in mines will depend on their relative strengths under these forces. The relative strengths vary in different mines and in different beds in the same mine. The tendency is to draw conclusions from the appearance of the immediately visible roof or top rock without considering what action may occur in the overlying rock mass.

No satisfactory results can be obtained by attempting to relate the mode of failure in small pieces of rock in a testing machine with what occurs in large masses of rock under stress in mines, such as, for instance, the formation of fractures due to advance of working face.

Results of reinforced column investigation at Lehigh University. Slater, Willis A., Eng. and Eng., October, 1931, pp. 219-228. In reinforced concrete columns the stresses in the reinforcement increase through flow of concrete. In general, columns that have a stress of 8,000 to 10,000 pounds per square inch at the beginning of loading may increase in stress to 42,000 pounds per square inch, with no indication of failure. Furthermore, tests show that columns with plain ends -- that is, with uniform bearing on concrete and ends of reinforcement bars -- have greater strength than columns with other arrangements of reinforcing bars.

The extensive use of concrete columns in mines makes this information of particular value to miners.

Strata movements induced by longwall workings. Winstanley, A., Iron and Coal Trades Rev., London, February 5, 1932, pp. 246-247. Results of investigations with subsidence recorders, strain gages, and so forth, have made it possible to secure continuous records of movements of roof, floor, and in beds particularly in advance of longwall faces. It is shown that the movements are similar in character but different in magnitude in each instance.

As the working face advances, the roof beds lower under the force of gravity, but resist it to a certain extent and lower most over the area where the bed is being removed, transmitting part of the force over the solid bed, which decreases in thickness and yields outward at the free face; a similar action occurs in the bottom rocks. The significance of the converging movements taking place in advance of the working face is that the beds, when exposed at the face, have been subjected to distortion, and consequently have lost some of their strength properties. Roof beds when broken, in settlement, no longer offer resistance to gravity but become practically dead weights to be carried by the unbroken beds and the supports employed. Furthermore, the unbroken beds tend to distribute the effects of local forces, and thus avoid local damage to other beds. It is desirable that the roof beds be kept in contact one with the other in order to prevent extensive breaks close to the face; the closer they are kept to the waste pack the better. The earlier and more effective the reinforcement of the roof beds by artificial supports, the greater the chance there is of preserving the beds, thus requiring less support.

Ground movement in mines. McTrusty, J. W., Iron and Coal Trades Rev., London, March 11, 1932, pp. 435-437. Any practical policy claiming to have a scientific basis must be preceded by an investigation of the fundamental factors involved, the problems of safety and economy requiring an understanding of the cause of movement and logically improved methods of support and methods of working.

There are three areas involved in ground movement: (1) The subsidence area, (2) the middle active-pressure area, and (3) the outer or dead area. The types of movement do not vary much in the three areas, but change with the geological conditions, methods and rates of working, and the support adopted. The separate movements are interrelated and produce complex movement. The formation of faults has its counterpart in fractures produced by movements resulting from mining. All rocks are compressible at depth to an extent determined by their constitution; the energy of compression stored within the beds adds something in the nature of "live" loads to the dead weight of the material beds, hence the reactions of the strata liberated by

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mining operations are not necessarily simple weight reactions. Time, pressure, and the conditions of restraint have to be taken into account when considering the flow or simulated flow of sedimentary rocks.

Fractures become inevitable whenever beds are strained beyond the resisting capacity of the material. Ordinarily the roof immediately above a bed at the working face is subjected to the weight of the superincumbent strata, and to compressive stresses from the cantilever action of the nether roof during subsidence; the presence or absence of natural fractures in the top rock must influence the concentration of stress.

Experience indicates that the safest roof conditions as well as the best working conditions are obtained where the roof over mined-out areas can be held for a considerable period close to its original position, which condition can be obtained to best advantage with systematic, straight-face working and adequate support. The success of any system of support is influenced, however, by the strength of the bottom rock on which it is erected.

The subsidence area in coal mines. Trans. Inst. Min. Eng., London, December, 1931, pp. 249-250. Subsidence is an inevitable consequence of extracting a seam of coal by the longwall method of working. The distance through which the roof subsides is roughly proportional to the thickness of the seam extracted, and, broadly, is about 50 per cent of the thickness when ordinary systems of packing are adopted.

Strong packs tend to reduce the rate at which the roof subsides, whereas weak packs permit an increased rate of subsidence. The force of the lowering roof mass within the subsidence area is practically irresistible, and ordinarily props are not suitable types of support for general use in roads within this area.

Surface damage through settlement due to mining. Booth, H. A., Coll. Guard., London, February 5, 1932, pp. 251-253. It is claimed by some that settlement of the surface following mining will cease in five years, but will depend upon the depth of the seam, the thickness of the seam, and the character of top rock. Furthermore, unmined pillars and props left in the workings delay the ultimate settlement. It is usually held that the surface may be affected to a maximum horizontal distance equal to one-third the depth of the seam; however, this may be influenced by proximity to old workings when the damaged area may be extended. The extent of damage is also affected by the character of the overlying beds and the dip of strata.

Rock bursts in the Mysore gold mines, India. Iron and Coal Trades Rev., London, April 8, 1932, p. 593. Rock bursts are of common occurrence in these mines and often occasion considerable damage to the workings, as levels and stopes, the former being more subject to such disturbances. Rock bursts occur without warning and with practically irresistible force, crushing large timbers and often demolishing several hundred feet of levels. The accompanying air blasts are serious and extensive, affecting workings for a mile or more.

The rock walls consist of a homogeneous gneiss, thousands of feet thick, which does not easily fracture and is not plastic. A theory advanced is that the walls are partially elastic and will yield to a very small extent on the removal of blocks of ground by mining, following which they remain stationary but continue to store up energy due to potential movement. Eventually the potential energy overcomes the resistance to fracture and the rock bursts.

No attempt has been made at scientific control of the roof in the mining of the lode, economic requirements being alone considered; and further, no support was employed in the upper levels, the effect of which on pressure in the lower levels is cumulative and adds greatly to the loads that are thrown upon such supports as are provided.

Rock burst in the Mysore gold mines. Min. Jour., April 9, 1932, p. 239. Much damage results from rock bursts in workings such as shafts, levels, stopes, and winzes. As a result of much investigation the workings have been made more secure and safe-guarded by the use of granite pack walls in stopes, and by the use of concrete and brick linings for levels and winzes. The value of such supports has been proved and their extension is steadily proceeding.

Pressure of rock masses in conglomerate lode. Vivian, Harry, Inf. Cir. 6526, Bureau of Mines, 1931, pp. 4 to 6. An interesting discussion of cause and effect of rock pressure in the deep workings of the conglomerate lode mines is given. The presence of joints, slips, and other breaks seem to deny the idea that unrelieved strains and stresses are the cause of violent breaking of pillars. Faults and slips contribute to the distribution and concentration of pressure in the workings, resulting in (1) a sudden increase of rock pressure upon certain pillars or temporary supports, (2) concentration of load at points on walls of fault, (3) shifting of loads on areas bordering caved ground to actively worked portions.

The weight of overlying broken rock acting on pillars within mine workings probably causes rock bursts, which will increase with depth, both in frequency and severity. The pressure block, consisting of broken rock above the stopes, determines the load upon the supports and consequently its amount and kind. The load on the temporary support is directly proportional to the time the support has been in place for a given position.

Roof support in bedded deposits. Walker, Arthur, Sci. and Art of Min., London, April 16, 1932, p. 308. Support should be given the top rock as soon as possible after the material mined is removed, in order to prevent separation of the roof beds. Such separation may in some instances extend over the bed mined for some distance. Temporary support is effective only in respect to the separated beds, while the beds above are capable of movement to the extent of the separation. The greater the extent of separation, the greater will be the impact of overlying beds should they be released through the extension of a break cutting the top rock. The separation of beds should be kept at a minimum in order to reduce the amount of such impact. The greatest rate of deflection is adjacent to the face, where the rate of separation is also greatest, for which reason the most effective supports are those set close to the face. Props set close to the face until space can be provided for more permanent supports check the deflection of the roof beds and materially aid support of the top rock.

Support of underground workings in Rand mines. Min. and Ind. Mag., South Africa, March 2, 1932, pp. 26-30. The dip of the roof varies from 5° to 20° on the far eastern portion and from 60° to 85° on the far western portion of the northern outcrop. The depth of mining operations is nearly 8,000 feet and support must be adequate for local needs, and also to maintain the integrity of adjoining mines. The types of supports employed comprise pillars, timber props, pigsties or cribs, mat packs or chocks, steel members, stone walls, concrete, masonry and brickwork, ore from shrinkage stoping, ore on stulls, waste from resuing, and sand filling. The method adopted depends upon the nature of the excavation, angle of dip, depth of workings, nature of the orebody (particularly the hanging wall), and thickness of the orebody. Props, pigsties, packs, and concrete, are used on dips of 0° to 35°; packs supported by pigsties or stulls, on dips of 35° to 45°; stulls against props for over 45°; stulls on skeleton pigsties or mat packs if the ground is heavy; the Randfontein method, which consists of spaced props, mat packs, and stulls for receiving the broken ore, placed at more or less regular intervals on dip and strike; and finally, shrinkage.

Increased rock pressure has made pillars unsatisfactory except as they may be employed in conjunction with other forms of support. Artificial supports have the advantage of being strong, tough, and durable. Timber is still the mainstay for underground support, the roughest kind of timber has been found adequate for props, pigsties, chocks, and various modifications. Steel has not been found desirable for support, while concrete, especially the concrete disk, is eminently suitable for certain dips and pressures. Stone packs commend themselves mainly on account of cheapness. The Sloan wire pack is a useful and often satisfactory type of support. Sand filling was formerly used mainly for strengthening pillars, but on account of its steadying effect it has been widely extended for ground support.

A method of preventing crush to brick stoppings and similar structures. Leeds, C. H., Trans. Inst. Min. Eng., vol. 82, pt. 4, pp. 310-311. Brick stopings or walls built in unsettled ground often fail because of roof pressure or creep, and the consequent effect upon maintenance of workings and derangement of ventilation may be disastrous. It is better to take precautionary measures than to rebuild supports.

It is claimed that stoppings may be built that will largely obviate failure and the expense of rebuilding. A foundation is prepared in the ordinary way and the brickwork built up to a height of 12 inches. On top of this wall, bricks are laid on edge flush with the face and back so as to form a trough 21 inches wide. The trough is then filled with peat moss, and upon which are placed undressed boards 27 inches long by $\frac{1}{2}$ inch thick by about 5 inches wide, forming a complete covering from end to end of the brickwork. Building of the brickwork is then resumed upon the top of the boards, the thickness of the stopping being now reduced to 18 inches so as to allow the bricks on edge supporting the ends of the boards to stand out clear of the brickwork above them. When brickwork of sufficient weight has been built over a portion of the length of the stopping, the moss is rammed beneath it in order to make

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the combined structure as solid as possible, and the stopping is completed to the roof and wedged in with slate or other suitable material. As the weight increases, the boards break at the edges of the upper portion of the stopping and allow it to settle down gradually on the moss.

Effect of loads on props. Iron and Coal Trades Rev., London, February 5, 1932, p. 246. Measurements made on dynamometer props show that they are frequently swung under the influence of movements in the roof and floor. This causes eccentric loading, and the props are bent and damaged under loads much less than they could sustain when applied axially. The conclusion drawn is that props set in the usual way must not be expected to give full control to the strata movements. The variable and varying resistances of the props cause frequent local changes in the movements of the roof, and the props themselves are liable to become unstable; they nevertheless contribute appreciably to the control of the rock movements.

Effect of failure of caps and penetration of beds by props. Trans. Inst. Min. Eng., London, vol. 82, pt. 4, p. 334. When a cap splits or a prop penetrates a floor bed, its resistance decreases, and the material of the prop recovers under the reduced stress, which tends to the preservation of the props; but the loss of resistance is bad for the roof and floor. With unyielding end-contact surfaces, the shortening of a composite steel prop or even of a wooden prop is only a small fraction of an inch to the elastic limit of the prop. The yield to the convergence of roof and floor is therefore taken by the crushing of caps or penetration of the floor.

H-section props, with the same contact surfaces at the ends as cylindrical props, are weaker to resist bending at right angles to the web. A modified form of H-section prop with the flanges turned over to form an almost cylindrical section has been suggested.

TIMBER AS SUPPORT

Timber supports in the workings on the Rand, South Africa. Min. and Ind. Mag., South Africa, March 2, 1932, p. 27. Timber in various forms superceded pillars as the workings increased in depth, the eucalyptus, acacia, and cypress being the woods employed. The simplest form of support is the prop, which while temporary in nature has the advantage that it serves as an indicator of weight of the hanging wall. Props are readily adapted to varying dips and thicknesses of reef. Headboards are generally employed at the top of props to serve in cushioning and distributing the weight.

Pigsties are still considered the most efficient and economical support for dips up to 50° and for all depths. As usually constructed they are 4-sided and are left empty or filled with rock; skeleton pigsties are favored for dips of 45° and over. To insure strength and compactness the timbers are notched near the ends.

Mat packs or chocks are built up of short lengths of timber, (about 2 feet, 8 inches) and are wedged in place; they are usually employed in narrow reefs. The "duplex mat" is a square mat pack formed by superimposing layers of four lagging holes, spaced 4 inches apart and held together by rods.

Rough timber is ordinarily used and has been found adequate, but occasionally squared timber is found necessary, particularly where joints are required.

Use of props in coal mines, England. Coll. Guard., London, April 1, 1932, p. 635. The use of props in coal mines differs but little from similar use in all bedded deposits, consequently the following suggestions are of interest: (1) Rules relative to maximum distances in placing props should be adhered to for best results; (2) as the props next to the face are probably the most important, great care should be taken in placing and spacing them; (3) caps should be prepared at the surface and be of uniform size; (4) props at the face should be staggered in order to decrease the area of the roof unsupported; (5) faces should be carefully and regularly inspected and points marked by chalk where props are needed; (6) steel props should be used whenever possible as face supports, as they offer an even resistance to the roof, and the probability of induced breaks is reduced; (7) steel caps (flats) should be employed particularly at points needing special support; (8) all supports should be removed from the gob in order to insure even subsidence of the roof.

Homegrown pit props. Coll. Guard., London, February 19, 1932, p. 346. The Forestry Commission, in England, is rapidly increasing the output of timber suitable for pit props, and much larger quantities will become available in the near future. There is a possibility that the output of homegrown Scots pine, larch, and Douglas fir for pit props can be raised to 500,000 tons a year and even more as the State plantation develops. The importance of uniformity is pointed out and the suggestion is made that care should be taken to avoid marketing props with tapered ends. The props should be standardized as to size and shape.

Cantilever roof protection. Queensland Govt. Min. Jour., November 14, 1931, p. 437. As a means of protecting men working under loose or broken ground, a cantilever of timbers is used which consists of two 6 by 6 inch booms, held in place below the caps of the last two sets next the face of a drift by four strap-iron hangers. The booms can be slid back and forth in the hangers and with pieces of plank placed upon them crosswise form a protective lagging for the men. During drilling the booms are removed or drawn back in order to give plenty of room, and during mucking they are advanced to the drift face. Little space is occupied by the timbers and they can be put in place quickly.

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Use of square-sets in the Butte copper mines. Harrer, C. M., Eng. and Min. Jour., March 1932; pp. 146-148. Square-sets and filling are commonly employed in the support of stopes in the Butte mines. The working face is carried up by offsets, and on the completion of an inclined slice, filling is introduced into the stope by merely knocking out or blasting out the sides of the end raise chutes facing the stope. The waste runs into the stope from the raise chute to its angle of repose.

The system of alternate stoping and filling has necessitated the division of the stope blocks between two raises into a double stope, one wing or side undergoing filling while the other is in the productive stage. Formerly, this method of mining was considered applicable only to steeply dipping orebodies and hard, firm, hanging wall; now stopes are worked with comparatively weak hanging walls by keeping up the filling and employing stulls.

Where the hanging wall is quite weak, a system of connected timbering, square-set, or stulls is employed as an auxiliary support. If the vein is rather wide, staggered square-set timbering is used. The inclined-cut-and-fill method has, therefore, a wide application at Butte.

Notes on timbering shafts in deep-seated placer deposits. Roberton, J. Hume, Canadian Min. Jour., October, 1931, p. 759. The size of timbers to be used in support of dirt, sand, and gravel deposits, usually saturated with water, is considered from the theoretical standpoint. It is estimated that for shafts 100 feet deep, timbers 14 inches wide by 15 inches deep would be necessary.

Use of timber for shaft lining. Eaton, Lucien, Eng. and Min. Jour., February, 1932, p. 85. The advantages of timber for shaft lining are (1) it is usually the cheapest and most convenient material to use, (2) it is easily framed and placed, (3) it is not eaten or corroded by acid water, (4) it gives warning before it fails, (5) when broken by pressure or accident, it can be chopped out and replaced with a minimum of difficulty or loss of time. Its chief disadvantages are (1) its bulk, for it takes up more room than steel, (2) the danger of fire, (3) its comparatively short life when it is not always wet.

Use of hanging bolts in shaft timbering. Eng. and Min. Jour., November 23, 1931, p. 452. The use of hanging bolts in supporting and holding cribbing in place in shaft sinking is described and sketches of them in use are given.

The preservative treatment of mine timbers. Vaughan, R. J., and Prettie, R. J., Canadian Min. Jour., October, 1931, p. 756. Wood has a wide range of use in mines, varying from temporary to permanent support, and an extensive application in secondary support -- that is, as an adjunct to all other means in maintaining mines. It is shown that failure of mine timbers in the mines of the United States is due to the following causes and is in the following proportions: (1) Failure due to decay and insects, 50 per cant; (2) failure due to breakage and fire, 20 per cent; (3) waste, 25 per cent; and (4) wear, 5 per cent.

Causes of decay are moisture and heat, especially under variable conditions of both; the most practicable means to be employed in preventing decay is by poisoning the food supply of the fungi -- that is, by the use of preservatives. An ideal preservative is one that is both toxic and volatile, the latter in order that it may be readily and efficaciously applied.

The paper deals with the use of zinc chloride and of creosote as timber preservatives, giving strengths used and method of application.

Bad air and timber decay. Min. Congress Jour., March, 1932, p. 26. It has been observed that when mines are closed for considerable periods, much expense is incurred in resumption of operations because of timber decay and the resultant caving of overlying rock. In certain instances it has been noticed that in the parts of mines in which there was good air circulation, there was a minimum of this decay of timber. The inference is that mechanical or controlled ventilation might pay if used only to protect timbers in both working and idle periods.

Portable timber saw. Coal Age, March, 1932, p. 128. A new type of the Wolf gas engine driven portable timber saw has been developed for cutting heavy timbers in the woods as well as for cutting and trimming mine timbers. These saws can now be obtained with any one of the three common power drives, as compressed air, electricity, and gasoline. Standard models have a capacity of 24 inches with a range of 18 to 48 inches. The gas engine used with the new type is of the aircraft design, being light and powerful.

FORMS OF METAL SUPPORT

Steel supports in the Rand mines, South Africa. Min. and Ind. Mag., South Africa, March 2, 1932, p. 28. Steel employed as support other than in shafts and as joists is not in favor in the Rand mines. When used as girders, or joists, the ends are inserted in hitches cut by machines in the solid rock of the walls of haulage ways. Often pigsties are built on top of the girders to support bad sections of the roof. The advantages of steel over timber are: (1) It is more durable and consequently cheaper to maintain; (2) it can be reused several times; (3) is lighter and easier handled than timber of equal strength; (4) takes less room than timber; (5) does not decay and pollute the air; and (6) is noninflammable. It is, however, subject to corrosion from acid mine water and tends to weaken structurally under alternating and prolonged localized stresses.

Application of steel props in general. Coll. Eng., London, February, 1932, p. 69. As a result of experience in the use of steel props in English coal mines, their value has been proved particularly under favorable conditions of a clean seam 30 inches or more thick, with a variation in height of roof not exceeding 2 inches and having no dirt band with which to contend. With slight variation in height of roof a definite length of prop can be used. The initial cost of supports prohibits indiscriminate loss if economical working is to be attained. Furthermore, successful roof control demands complete withdrawal of all steel supports in the waste, It is therefore essential that a detailed record of props be kept.

Use of steel props in mines (England). Sci. and Art of Min., January 23, 1932, p. 210. The use of steel props is growing in the English coal mines, the forms in most favor being the H-section and the tubular types. The advantages of the use of steel props over wood are given as follows: (1) The height at the face is maintained owing to the rigidity of the props; (2) the roof breaks at the face are less because the props do not allow the roof to lower and fracture; (3) the stronger support that they affort ensures that the faces will be kept open; (4) they save the setting and withdrawal of wooden checks; (5) by being of uniform strength they prevent the formation of cross breaks; (6) better control of the roof is assured, giving improved working conditions; (7) there are fewer falls and a greater factor of safety; (8) less time is lost at the coal face, and consequently provide more regular work and better earnings for miners; (9) a more definite line of roof break is obtained; (10) minor coal face accidents are reduced, thereby obviating loss of time by miners.

In one instance where 6,500 steel props were used there was a loss of only 2 per cent per month, indicating that four years is the life of a prop.

Use of steel props in English coal mines. Luty, B. E. V., Iron and Coal Trades Rev., January 22, 1932, p. 137. Steel props are finding increasing favor, particularly the solid type and tubular form, the latter being considerably lighter than the girder prop of the same strength. It is also a fact that a considerable amount of squeeze can be taken without distortion of the tube; and when the wood ends are too badly mopped up for further use, a portion can be bored out and new end pieces inserted butting on the core so that the prop is again ready for use. Wooden cores should always be provided when tubular props are employed; while they may reduce the possibility of distortion, their real function is to simplify the reconditioning of the tubes.

Use of steel props in coal mines. Iron and Coal Trades Rev., January 29, 1932, p. 210. Owing to the extended use of steel props in English coal mines, it is interesting to note certain conditions under which they are not applicable. When the roof is weak and it is desirable that the weight should act slowly, the steel props resist it, with the result that they are pushed up through the roof and have to be abandoned.

Steel supports for roadways and steel props for face work. Trans. Inst. Min. Eng., London, October, 1931, p. 9. Failure of arch girders in roads is due to the following causes: (1) Improper packing of the crown of the arch; (2) side pressure forcing the crown upward and breaking the fishplates; (3) bad design of parts such as fishplates or too light arches; (4) improper spacing of arches; (5) imporper bracing of arches.

Lack of success in the use of steel props is probably due to use with other props rather than inherent defects in the props, themselves. Do not mix steel with wooden props.

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Steel props and arches for roof support. Clements, Fred, Iron and Coal Trades Rev., London, January 1, 1932, p. 5. Owing to the nature of the stresses imposed upon props and arches used for roof support, it is imperative that the steel used should be (1) absolutely reliable, uniform and free from surface and internal defects, and (2) possessed of a high capacity for undergoing bending without fracture. Steel supports are subject to bending, which often ultimately and inevitably exceeds the yielding point of steel. Other things being equal, the capacity for undergoing bending without fracture is at a maximum in low-carbon steels, and decreases as the carbon content rises; therefore open-hearth steel is definitely superior to other kinds.

The use of camber girders. Trans. Inst. of Min. Eng., London, December, 1931, pp. 259-260. A camber girder consists of a suitable length of girder, curved in its middle portion but having a part of each end straight and tangential to the curved part. The camber is used in lengths of 5 to 12 feet and varies from 12 to 30 inches in the height of camber. The greatest height is generally used in openings made in weak top rock, but as a result of experience there is a tendency to reduce the height in proportion to the length of the girder. For instance, girders 11 feet long with a camber of 12 inches have proved successful in hard rock.

When the wall rock is reasonably strong, camber girders are secured in position by wedging the ends to strong wooden blocks placed in pockets cut in opposite sides of a passage, the blocks being set at right angles to the axis of the girder. Cast-steel shoes are sometimes used. In weak ground girders may be grouped in threes or fours and wedged on wooden bars or sills. Ordinarily girders are spaced 3 to 4 feet apart, uniform spacing being best.

Use of cambered girders in mines (England). Sci. and Art of Min., January 23, 1932, p. 210. The use of camber girders or "springs" in English coal mines is common practice, particularly where there is rock movement or in "motive" zones near the coal face. Camber girders are simply girders bent to a curve and set from the walls instead of the floor. They take the place of the old-fashioned, but effective, herringbone strutting, but being of steel and without joints, are very much stronger. They allow the road to settle naturally with the adjacent strata and at the same time support the roof and opposing side squeeze.

Use of cambered girders in subsiding area. Trans. Inst. of Min. Eng., London, vol. 82, p. 344. As settlement of roof takes place the cambered girder settles also, whereas the arch girder pushes up into the roof, often breaking what would otherwise be a sound top. It is much easier to make repairs by taking up the floor than to remove arch girders and reset them after reripping.

Use of steel arches in mines (England). Sci. and Art of Min., January 23, 1932, p. 210. The use of "horseshoe" arches of rolled steel, commonly known as "rings" are coming in general use, one mining division having 73 miles of roadway supported by them, while another division had about 350 miles of roadways supported by steel rings in 1929.

Arches bent and distorted by rock pressure can be straightened and put in use again with apparently as good results as when new. The withdrawal is also readily accomplished, as is shown by an instance where 6-foot arches were withdrawn from a roadway 600 yards long at the rate of 12 arches per shift of two men.

An E arch with its accessories consists of 2 half arches, 2 fishplates, 4 bolts and nuts, 36 struts of round timber, 6 notched struts, 2 sole pieces, 2 clips with bolts and washers.

The support of junctions by steel arches. Fowkes, Alfred, Iron and Coal Trades Rev., London, February 5, 1932, pp. 233-234. Steel arches have been found very satisfactory for the support of ground which had formerly proved difficult when braced with girders and props; however, their use has been confined largely to coal mines in England and on the continent. The concentration of weight at the junction of roadways causes trouble due to change in sequence of supports, and distortion of the girdered junctions occurrs. Various types of steel structures were tried before the present form of support was adopted. Arches made of 8-foot steel are now carried as close to the edge of the junction as possible and two pairs of the ordinary standard 12-foot girders are bridged diagonally across the corners of the junction, one pair passing under the other; the legs of the 12-foot arches abut closely to the 8-foot arches, the whole of the arches forming a complete anchorage with each other and present the appearance of a vaulted ceiling supported by columns.

In order to get the diagonal arches, one above the other, it is necessary either (1) to set one pair of diagonals upon an extra foot block or (2) to cut approximately 6 inches off the ends of the legs of one pair of diagonals, the second method has proved the more successful, as it keeps the whole system of arching uniform.

The support of underground roads by steel arches. Coll. Guard., November 27, 1931, pp. 1790-1792; December 4, 1931, pp. 1880-1882. The use of steel arches for the support of mine roads in the Yorkshire coal mines is being rapidly extended. Considerable skill is required to erect the arches due to a wide difference in character of the roof, floor, and sides of the roadways. Three types of arches are in use - horseshoe, straight-sided, and splay-legged-- which generally conform to standard sizes. Details are given as to size and weight of the various types used. Problems that require special consideration and study in the application of the arches are: (1) Subsidence; (2) the development of pressure upon roof and side supports; (3) stability of roof and sides so long as supports are not disturbed. The special conditions under which the types of arches are applicable are outlined. The protective devices employed with steel arches are: (1) Lateral support between arches to prevent sidewise buckling; (2) lagging and packing to prevent radial distortion in the plane of the arch; (3) foot blocks to provide a yielding foundation; (4) correct spacing of arches.

The use of steel in shaft lining. Eaton, Lucien, Eng. and Min. Jour., February 1932, p. 86. The advantages of steel as a material for shaft sets are as follows: (1) It takes less room than wood; (2) it is strong, bends more often than it breaks, when bent can often be straightened on surface and can be used over again; (3) it is easily placed; (4) it is fireproof; (5) it does not rot and is not subject to the attack of insects. Its disadvantages are: (1) It often costs more than wood; (2) when wrecked through accident or excessive pressure, it can not be chopped out, and it can not be cut to fit underground; (3) it is corroded by acid water.

Corrosion of iron and steel. Iron and Steel of Canada, December, 1931, p. 186. The American Society for Testing Materials has recently issued a report by Committee A-5, on the corrosion of iron and steel, which gives the result of extensive tests of metallic coatings and inspections of galvanized sheets, hardware, structural shapes, conduits, and so forth, showing failure of a number of coatings. This work should be of interest to the mining engineer who has to do with iron and steel structures both on the surface and underground.

USE OF CONCRETE AS SUPPORT

Concrete supports in the Rand mines, South Africa. Min. and Ind. Mag., South Africa, March 2, 1932, p. 29. Concrete is usually stronger than the rock or ore in a mine and where properly placed constitutes a strong support. Concrete was first used in the mines as monolithic columns and later as built-up columns, "disks" or "pancakes" of concrete being superimposed to form the support. The disks have the advantage that they can be made on the surface. When the columns have a soft formation as a footing and have a capping of timber they give remarkable good service.

The disks range from 27 to 33 inches in diameter and are 3 to 4½ inches thick, with a central 4-inch hole. They are reinforced by wire rope. The concrete mixture is 1 of cement to 3 of sand and 6 of 1½-inch-mesh rock. Uniform mixing and treatment in a "pickle" bath for a week insure excellent disks.

The advantages of concrete over pigsties are: (1) Less risk of fire, (2) they become harder and stronger with age, (3) stopes can be kept cleaner, (4) they are easily erected, and (5) supervision is made easy.

The Schaefer lining for roadway. Coll. Guard., London, April 15, 1932, pp. 734-735. The Schaefer concrete lining has been employed in the English coal mines, although up to the present time to a rather limited extent. The lining consists of a series of concrete arches abutting one against another with or without inverts and suited to the particular conditions. The arches are constructed of well-matured concrete blocks in the form of the letter T, and with a proper taper to permit close contact of the faces, as they are built dry. Through holes cast in the rib of the blocks, steel rope is passed as reinforcement, and is secured to the blocks by grouting as the lining is erected. The invert is first constructed and the arch is built on steel centers which are advanced as the rings are completed. Wooden compression courses are inserted where pressure is expected to develop.

Larger excavations are required for these linings than with most other methods of support, and they are more difficult to erect. Furthermore, their success as elements of support depends largely on the careful packing of a considerable quantity of small-sized waste back of the lining. Continuous Schaefer lining is costly, for which reason the rings forming the lining should be spaced, thus making it comparative with steel arching. One of the merits claimed for the Schaefer lining is that each ring can move independently.

Use of concrete for shaft lining. Eaton, Lucien, Eng. and Min. Jour., February 1932, pp. 86-87. The use of concrete as a lining for shafts has increased rapidly, owing to strength and increased facility in handling and placing. The advantages of concrete are as follows: (1) It is fireproof; (2) its strength increases with age; (3) it does not rot, nor is it corroded or easily disintegrated by acid water, although this does sometimes occur; (4) it may be poured in place, making a continuous support; and (5) it may be used as precast blocks, which afford greater uniformity in strength.

Cementation in shaft sinking. Detchon, R. J., Sci. and Art of Min., January 9, 1932, pp. 194-195. The author describes a method of cementation used in shaft sinking that came under his observation, where water occurred under pressure. Boreholes were put down near the edge of the shaft and as truly vertical as possible, approximately 100 feet deep. The boreholes, four in number and 2½ inches in diameter, formed a square within the shaft, being an equal distance apart and within 2 feet of the shaft sides. Short lengths of 2-inch pipe, 2 feet 6 inches to 3 feet long, were inserted in the boreholes with threaded ends upward, and wedged tight with soft wood plugs or wedges, thus sealing the water off and allowing it to discharge through the pipes into the shaft. Valves were fitted to the ends of the pipes and the pipes connected with the pumps circulating the cement mixture.

The cement was mixed in a barrel in the proportion of 95 parts of water to 5 parts of cement, or a ratio of 19:1, to form an intimately mixed liquid cement. The pump used in handling the cement had a 9-inch air cylinder, a 12-inch stroke, and a 2-inch duplex plunger. It was capable of developing a pressure of 1,600 pounds per square inch. At shallow depths the pressure of cementation was kept below that due to the head of water at the depth of the shaft, in order to prevent loss of cement through escape in fissures.

Tunnel grouting at Cobble Mountain. Hatch, Harry H., Eng. News-Rec., December 31, 1931, pp. 1037-1039. Voids formed or left behind concrete linings in rock tunnels require grouting to fill them and insure good contact and support. The maximum opening occurs at the crown and narrows to a minimum at the spring line; there are also irregular open spaces in the rock face. Drain pipes were placed at water seams and other sources of water, and drains were also placed in the lining to take care of running water. Grout holes were placed after the lining was complete, and were 2 inches in diameter and fitted with $1\frac{1}{2}$-inch pipe. Three types of grouting were employed according to placing of holes: (1) A single line of holes spaced at various intervals in the roof of the tunnel; (2) a hole placed on each haunch of the arch and located 30° above the spring line; (3) eight holes drilled 10 feet through the lining and into the rock, at 45° intervals around the section, beginning at the crown.

The grouting machine consisted of a small cylindrical tank with a flap gate at the top for charging materials; air was admitted at the bottom. A maximum air pressure of 100 pounds per square inch was used in placing the grouting. Following the initial charge of neat grout, more sand was added, but never more than 2 parts of sand to 1 of cement. Other details are given relative to practice under variable conditions, amount of grout used, movement of grout, check on filling of voids, and so forth. Costs are also given.

Cementation of cast-iron tubbing at Shireoaks colliery. Walters, L. Ward, Trans. Inst. of Min. Eng., London, vol. 82, part 4, pp. 320-331. A detailed description with sketches and photographs is given of cementation of corroded shaft tubbing in an English colliery. The tubbing was lined with 3-inch brickwork which adhering to the defective metal tubbing and caused openings to form when the brick was removed in the search for leaks and in preparation for cementation. Extensive breaches would cause a rush of water and reverse the ventilating currents. Furthermore, owing to the weakened condition of the tubbing, no great pressure could be put upon it. It was then decided to cement one section of the tubbing at a time, and ultimately to complete the whole length so as to form a continuous lining.

In order to prevent any excessive pressure on the tubbing, a T was put in the pressure main between the cement pump and the shaft. From this a vertical pipe was run for 15 feet and then carried back across to the mixing tank. When cement would not enter the tubbing, the pump would force it back into the mixing tank; the only extra pressure then imposed would be that due to the 15-foot head. The tubbing was tapped by drilling through the original plug holes, so that $1\frac{1}{4}$-inch nipples and provided with valves could be inserted. Flexible connections joined the nipples with the pump discharge circulating the cement mixture. No serious difficulty was experienced in the work, which was done satisfactorily and with no enforced delay.

The use of gunite in fractured and loose ground. Smith, T. E., Eng. and Min. Jour., March, 1932, p. 153. Gunite has been used successfully in various mines to hold together fractured and loose ground. The author describes the methods employed in the Butte mines which have proved satisfactory. When retimbering drifts through loose, blocky ground, gunite is used to prevent breaking down and running. The side lagging is removed and gunite is distributed over the exposed rock surface and in the cracks and interstices that can be reached. After three or four hours, the back lagging is removed and the loose material is cleared away. After the removal of back lagging and loose ground, all cracks are filled with a stiff mixture and all exposed rock surfaces are covered with 1 inch of gunite. A stiff gunite is more suitable than a thin gunite for cracks that can be reached readily, while thin gunite will penetrate gractured ground that is more inaccessible. A relatively dry mixture should be used for coating exposed rock surfaces.

Gunite filling will steady loose ground better than numerous sprags and stulls; also, an important advantage in the use of gunite is the warning given by formation of cracks.

About three cars of gunite material, including 1,000 pounds of sand and three bags of cement, are required per set, two men performing the work of changing sets in two days' time.

Underground air-drying pockets. Alan Wood mine, Dover, N. J., Min. and Met., November, 1931, p. 492. A description is given of the construction of an underground ore pocket for handling wet ore prior to its delivery to the mill. The cementation of the rock walls in order to keep out water was accomplished by the François method. A lining from 5 to 10 feet thick was provided without decreasing the size of the pocket as cut in the rock; that is, the lining was formed by grouting the rock walls. The cost of cementation was reduced by two-thirds and the time required by one-half. The rock is gneissoid granite.

USE OF WASTE PACKS AND FILLING

Packing waste in coal mines. Brass, T. F. S., Trans. Inst. of Min. Eng., London, November, 1931, pp. 149-152. The support of underground workings by the use of waste filling is of such importance that information relative to its application in all mines is of interest to the mining engineer. The methods of packing waste in the Continental coal field in Europe are as follows: (1) Hand-packing; (2) pneumatic packing; (3) blowing the dirt from a conveyor into the gob; (4) dummy-road packing; and (5) partial packing. The source of the packing material is from driving, ripping, dirt in seams, washing plants, and surface refuse materials.

Hand packing is costly in transport and placing. It is delivered by cars, conveyors, and chutes and put in place by hand. If coarse, it is built into walls, whereas the while small material is confined in inclosures formed by wire netting. Pneumatic packing is under 3 inches in size and is discharged through bunker and pipe under air pressure of 3 to 8 pounds per square inch. The packing is conveyed about 300 yards, beginning at the foot and advancing upward by removing sections of pipe. Eight yards are packed for each position. About 200 cubic feet of free air are used per cubic foot packed. When packing is blown in place, it is fed to a conveyor, from which it is discharged onto an inclined plate and thence fed into a tube and blown into the gob. No pack walls are built, as the material is stowed in a damp condition. Dummy-road packing is done in beds under 36 inches in thickness, the dirt being obtained by cutting dummy gates in the face and packing it solid between the gates. In the partial-packing method, chocks or packs of fallen waste are built across the mouth of each room and are reset as the face is advanced. All timber is withdrawn from the gob, the packs and fallen top rock forming the support.

Stone packs in the Rand mines, South Africa. Min. and Ind. Mag., South Africa, March 2, 1932, p. 29. Stone packs are cheap, as the material used is at hand; even ore may be used to be reclaimed later. The outer walls of the packs are strongly and evenly built to prevent collapse under pressure of the hanging wall. The walls may be reinforced by old rails, pipes, and timber laid crosswise. Packs are effective for dips up to 35°; they are built from below upward, are supported on the lower side by poles, and may serve to fill in between other supports such as concrete and pigsties.

The Sloan wire pack has proved very effective under certain conditions, the advantages being strength, lightness, the ability to hold the contained rock under compression, and cheapness of construction. The pack is made of 6-inch mesh wire netting which incloses a mass of rock or may encircle a pillar that is failing and reinforced by rock filling. The more the netting is compressed and reduced in width, the stronger the pack becomes.

Rock bursts in coal mines and effect of stowing. Bracht, _, Marx, _., and Spackeler, _., Ztschr. Berg-Hütten und Salinenwesen; vol. 79, 1931; Coll. Guard., January 8, 1932, p. 72. All rock bursts present certain concurrent and characteristic features, and may therefore be assumed to have a common cause. There are two kinds of bursts - stress and roof. The former are caused by differences in compressive stresses as governed by depth and nature of workings. Heavy roof bursts are caused by flexure of the solid main roof, leading to sudden breakage, under the considerable forces released by elastic deformation. The shock is transmitted to the rock adjacent to the working face, but at a considerable distance from the seat of the disturbance a wide variety of reactions may occur.

Stowing has frequently been suggested as a remedy for rock bursts, but has not proved satisfactory. Stowing can not prevent solid roof strata from bending or sagging up to the dangerous fracture zone, but it can retard the effect. The result of stowing may be that roof bursts will occur more seldom, but with greater intensity.

Disadvantages in the use of waste filling. Bull. Inst. Min. and Met., London, March, 1932, p. 4. Some of the disadvantages in the use of a filling of waste-rock ranging in size from 8 to 10 inches (about 80 pounds in weight) to plus 1/8-inch, are given for one instance as follows: (1) The stopes had to be timbered as worked, the ground being heavy though hard; (2) all timber had to be removed as filling was placed in order to avoid fires; (3) a minimum of dilution of the waste rock was permitted; (4) water admitted to the mine soon became acid and was costly to pump; (5) the ore beneath the waste filling had to be removed up to the filling overhead.

The rock used as filling, in this instance, was siliceous porphyry, and had to be loaded from shafts, trammed, dumped, built into walls, and shoveled into place. The specific gravity of the ore was about 4.8; that is, every cubic meter of void made produced 4.8 tons of ore theoretically. Owing to settlement of the ore only 3.25 to 4.25 tons were obtained for each cubic meter of filling placed. The amount of filling placed per shift varied from $1\frac{3}{4}$ to $3\frac{1}{2}$ cubic meters.

Support in the Mysore gold mines, India. Iron and Coal Trades Rev., April 8, 1932, p. 593. Owing to the variable width of lode, 1 inch to 30 feet, and the variable dip, horizontal to vertical, but usually about 65°, props and pack walls are commonly employed. Props are set securely into the walls, 10 to 12 feet below the level and usually three to a stope. On the props are placed poles forming a platform upon which is built the foundation wall about 2 feet thick. Side walls are next built up from the bottom wall to the level above, the inclosed space being filled with small-sized waste rock.

In order to hold the side walls in place, small wedge-shaped pieces are driven into spaces between the rock foot and hanging walls and the pack walls, thus fastening them in place so that the supporting props may be removed. The pack walls are thus built in sections from above downward and all timber is removed. As stoping is continued along the lode, other packs are erected at varying intervals. The worked out stopes are in this manner kept open for the passage of men and air currents.

Sand filling in the Rand mines, South Africa. Min. and Ind. Mag., South Africa, March 2, 1932, pp. 29-30. The use of sand filling has advanced from a minor rôle, such as that of supplementing pillars, to a major and standard form of support in the Rand mines. The sand is made into a light pulp by adding water and is then delivered by pumps to dewatering cones placed over the boreholes, through which it is conveyed to the mine workings. The cones return the overflow water to the sand tanks and separate out a certain amount of sand that it is desirable to keep out of the mines. Where inclined shafts are available, the sand is conveyed underground in launders. Where boreholes are employed they are usually 7 to 11 inches in diameter.

It is desirable to have a number of feeding points for proper distribution underground. Occasionally the sand has to be pumped to the boreholes and for considerable distances, often necessitating two lifts, owing to rising ground. Sand to the amount of 2,000 to 3,000 tons is placed underground daily by such a system.

In sand filling, a dip of 12° barely serves to move the material from the dewatering cones when 30 per cent of moisture is present. With a lower inclination more water must be added. A minimum amount of water should be used, as a surplus adds to the danger of loss of control and robs the sand of its supporting power. A three-walled chamber is erected in a stope at the location of a borehole, and from the down-dip side a launder conveys the sand to the points desired. Particular care must be taken in the preparation of an area for sand filling. Wire netting and cocoanut matting are often used to aid in strengthening and rendering tight the confining barricades.

The blast system of goaf stowage. Deuschl, E., Glückauf, July 4 and 11, 1931; Coll. Guard., November 6, 1931, p. 1537. The best high-pressure process is that developed by the Torkret Gesellschaft, in which the waste is fed into an upper chamber completely sealed off from a lower one. From the lower chamber the material is fed into the compressed-air stream by means of a bucket wheel rotating on a vertical axis. A pipe conveys the waste to the storage place.

The "cell-wheel" stower, also largely used, consists of a cylindrical casing in which a 6-cell wheel revolves on a horizontal axis. The cell-wheel serves as the distributor to the air main, the waste passing through it to the pipe below.

The main difference in the two systems is that the bucket wheel is subjected to a uniform pressure on all sides, while in the cell-wheel system the air pressure acts upward only. These machines are extensively used and handle the waste material successfully.

Pneumatic stowage of goaf. Poole, Granville, and Whitton, J. T., Mech. Handling, November, 1931, pp. 347-353 (reprint from Coll. Eng.). The objections to hydraulic stowage are partly overcome by the use of pneumatic stowage; they are as follows: (1) It is impossible to fill the void completely; (2) stowage in flat seams is more difficult than in an inclined seam; (3) drainage is made more difficult, including clarification and pumping of water; and (4) the effect on bottom and walls of workings is injurious.

Results of experiments with sand, sand and washery débris, and washing débris alone are given, with advantages and disadvantages of each.

Subsidence with different methods of filling. Mech. Handling, November, 1931, pp. 347-348. The following comparative figures give the effect on subsidence of the surface when using different methods and kinds of filling: (1) Hand stowage with material from 0 to 12 inches, subsidence 40 to 50 per cent; (2) mechanical stowage with material from 0 to 3 inches, subsidence 15 per cent; (3) pneumatic stowage at a pressure of 3 to 6 pounds per square inch, subsidence 25 per cent; (4) pneumatic stowage at a pressure of 15 to 22.5 pounds per square inch, subsidence 15 per cent.

Sand-filling methods at Hodbarrow (haematite) mines, South Cumberland. Jones, A. Alec, Bull. Inst. Min. and Met. Eng., London, February, 1932, pp. 1-1 The use of sandfilling of workings was to prevent water and water-laden sediments from entering the workings. The filling material is sand, readily accessible in unlimited quantities on the surface. The means of conveying the sand underground was by shafts and boreholes. At the beginning of the work the sand was sluiced into the boreholes, the pipes lining the boreholes being cut off from time to time to permit ready movement of sand and water. A bunker or pit was excavated at the shaft location into which sand was dumped and from which it was sluiced into the mine.

Launders were first used for distributing sand underground, but the limit was soon reached for this form of gravity filling, as there was not sufficient inclination for the launders. The next development was to make use of the momentum of the descent in boreholes and shaft. Cast-iron pipes 4 inches in diameter were employed, fitted with Victaulic joints and with a gradient of 1° from the horizontal. While there was a decided improvement in radius of operation, the volume of sand delivered was not improved, so that filling could not keep pace with ore extraction.

Compressed-air agitation was then tried and adopted. Compressed air was introduced into the delivery line at a pressure of 60 to 70 pounds per square inch. The injector finally adopted minimized the sand blasting of the pipe by injecting the air at a bend and along the axis of the delivery pipe. Injectors were also introduced at points in the delivery pipe in order to aërate the mixture and prevent choking of pipes. Furthermore, by aërating the feed a mixture containing a minimum of water is possible, the feed having less than 50 per cent of sand, by volume; up-grade deliveries are then possible. The longest pipe line was 300 yards and had many bends, with up-gradients of about 1 in 100, with three injectors more or less evenly spaced.

Continental mines employing "hydraulic stowing." Taken from paper on "Sand-Filling Methods at Hodbarrow, South Cumberland." Jones, A. A., Bull. Inst. Min. and Met., London, February, 1932, p. 19.

The following mines employ hydraulic stowing: Petite Rosselle, Forbach, Alsace, using a mixture of sand, burnt slate, boiler ash, and flue dust.

Velsen mine, Saarbruck, using the product of a sandstone quarry, the pieces not exceeding 2 inches in diameter.

Zwickau, German Silesia, using sand from a sand pit mixed with tailings.

The Concordia pit, German Silesia, using granulated stag mixed with sand.

Mystowitz, Poland, using sand excavated from sand pit.

Villeboeuf mine, St. Etienne, France, using a mixture of crushed conglomerate rock, cinders, with mine and washery refuse.

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Among others, the following mines also use hydraulic stowing in some form or another: The Ludvigsgluck Colliery of Upper Silica; the Salzer-Neuack Colliery of Essen, at Centre de Jumet, St. Louis, Belgium; and the Gewerkschaft mine of Thyssen.

Sand filling through pipes. Eaton, Lucien, Eng. and Min. Jour., March, 1932, pp. 137-138. Sand has decided advantages in filling for support in mine workings, such as slight compressibility and ease and economy of placing. The relative compressibility of sand and coarse material is 5 and 10 to 25 per cent. The finer the material used, the more complete the fill in underground excavations.

The methods employed in sand filling are: (1) By hand; (2) by water; (3) by compressed air, and (4) by air and water combined. When filling is done by hand, the sand is transported in the dry state in cars, dumped in raises, and distributed by cars, wheelbarrows, or scrapers. When water is used in handling and distributing sand, it is flushed down pipes or through boreholes, and distributed by pipes or launders. The material distributed may range from 20 to 75 per cent in solids. When compressed air is employed, the dry sand is introduced into the mine through pipes and distributed by an intermittent, high-pressure system through a receiving tank and discharge pipes, or by a continuous low-pressure system, in which the sand is fed into a pipe with a large volume of air. When compressed air and water are combined, the sand is flushed through pipes or boreholes and distributed through pipes with small jets of air introduced at intervals.

The advantages and disadvantages of the various methods are given with relative economies in their use.

Cement-gun control of loose and moving ground. Smith, T. E., Eng. and Min. Jour., March, 1932, pp. 152-153. The use of monolithic concrete structures have not proved successful as they crack and break up when subjected to pressure. In the Butte mines it was found necessary to gunite the failing concrete stoppings every two weeks. A method developed in 1930 to use sand as a filling for dams and stoppings has proved highly successful, and was first employed to isolate abandoned areas in the upper levels of the Leonard mine.

A bulkhead is first built with old timber across the drift to be closed, and after being wedged tightly to the sides and back, the loose ground all around the drift is removed. A second bulkhead or dam is erected 6 feet back of the first and to a height of 3 or 4 feet. The space between the two is then filled with sand, which is packed with water while being placed. The sand may be placed with a cement gun, which packs it tightly. The inner bulkhead is gradually raised until it nears the top of the drift, when the sand is shot through a narrow opening at the top. Any movement of the rock tends to pack the sand more completely.

When delivering sand through a cement gun, variations of pressure, rate of feeding, or flow of water are not important. By moving the nozzle steadily to and fro, the sand can be placed in a homogeneous mass. When delivered by a cement gun, the sand will stand vertically if 20 per cent or more of the material will pass a 30-mesh screen, and if the correct pressure and amount of water is used.

Sand filling on the Rand. Bull. Trans. Inst. Min. and Met., London, March, 1932, pp. 8-10. Sand filling as employed on the Rand is not to support the top rock but to enable the ore to be taken out completely. The method of working is by a kind of stall-and-pillar system. The dip is slight, being close to 14° to 15°, and pillars are left to support the roof, but as they are valuable it is desirable that they be mined. With this end in view, large areas are filled subsequent to extraction. The filling does not go on contemporaneously with the mining, but afterwards the mined areas are filled on a large scale, and then the pillars can be removed. In certain instances, to get at the pillars, it has been necessary actually to dig through the sand which has become consolidated by the great pressure to which it is subjected.

In the New Modderfontein the stopes are wide and the dip is slight. The upper portion of the mine was near the surface, and support was required to prevent caving, which would have admitted surface water to the mine. Hundreds of thousands of tons of sand had been lowered into the mine through boreholes and packed much better than other forms of filling which were tried.

The advantages of hydraulic sand filling in metal mines. Bull. Inst. Min. and Met., London, March, 1932, p. 7. The use of sand filling is of such importance as an aid to mining that its rapid extension is ensured where conditions are favorable to its application. The more important of the advantages are: (1) The possibility of completely filling large areas, as compared with the relatively small areas which can be packed with dry-filling; (2) the possibility of reaching areas which are partly caved and inaccessible; (3) additional safety to underground workers when the fill is carried out in current workings; and (4) the improvement of ventilation owing to old workings being completely shut off, and the prevention of fire owing to old timber being buried.

Rubber pipe lining (for filling stopes). Homer, D. D., Eng. and Min. Jour., October 26, 1931, pp. 367-368. Rubber-lined pipe for handling and distributing mill tailing as filling for stopes is described. Standard steel pipe 3 inches in diameter and 12 feet long is used, in which is placed a $\frac{1}{4}$-inch thick, vulcanized rubber lining. Special attention must be given to the connections; otherwise the sand will wear a passage between the lining and the pipe.

A 3-inch pipe has a capacity of 35 tons per hour, the cost per ton of sand being about 20 cents.